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MINING METHODS AND COSTS AT THE BRADEN COPPER CO.'S MINES, SEWELL, CHILE

BY

J. S. WEBB AND T. W. SKINNER

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MINING METHODS AND COSTS AT THE BRADEN COPPER CO.'S MINES, SEWELL, CHILE[1]

By J. S. Webb[2] and T. W. Skinner[3]

INTRODUCTION

This is one of a series of papers dealing with mining methods, practice, and costs being published by the United States Bureau of Mines. In this paper a description of the mining practice in vogue at the Braden unit of the Kennecott Copper Corporation is undertaken.

PRODUCTION

At present (1931) the entire production of ore is being obtained from the Teniente mine. The mining method in use is an adaptation of block undercutting which has been a development of the stoping and pillar caving experience obtained at this property. Improvements in mining practice are constantly being made.

GEOLOGY

The Braden Copper Co.'s property is located in the western cordillera of Chile, 30 miles northeast of Rancagua, 50 miles southeast of Santiago, near the head of the Teniente River, and at an elevation of 7,458 to 9,960 feet.

The western cordillera of the Andes forms a sloping dissected plateau. Near the mines the canyons are deeply trenched with long débris slopes and above these tower precipitous cliffs.

The mines are situated on the steep slopes south of the river, the lowest tunnel being at the bottom of the valley and the uppermost 2,500 feet above that level.

The orebodies lie around the periphery of an explosive vent in the form of crescent-shaped deposits, limited on the inside by the tuff contact, where they are of higher tenor.

The upper limit is formed by the bottom of the oxidized zone, 50 to 100 meters below the surface, and the lower limit by the contact with the primary zone.

The explosive eruption which formed the Braden vent intensely fractured the andesite porphyry surrounding its periphery. The width of the zone of most intense shattering is uneven, ranging from 100 meters to 600 meters, the widest portions being on the northeast side of the crater.

After an interval in which the crater became filled with bedded tuffs, mineralizing solutions rising about the periphery of the old vent deposited quartz, tourmaline, biotite, pyrite, and chalcopyrite in the irregular fractures in the andesite porphyry and formed

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2 One of the consulting engineers, U. S. Bureau of Mines, and mine superintendent, Braden Copper Co.

3 One of the consulting engineers, U. S. Bureau of Mines, and chief mine engineer, Braden Copper Co.

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large bodies of mineralized material with a copper content in most places of between 0.50 and 1.50 per cent, though locally slightly richer. This was the original source of the Braden ores, but their enrichment by descending secondary mineralization was necessary to raise their tenor to a commercial grade.

In the orebody pyrite increases and chalcopyrite decreases in relative abundance outward in all directions from the crater and the limit of ore of a profitable grade is generally coincident with the transition from predominant chalcopyrite to predominant pyrite.

The principal primary ore was originally chalcopyrite, but after secondary enrichment the mineralization is divided between chalcopyrite and chalcocite. The development of chalcocite is greatest in the upper part of the enriched zone, and gradually decreases with depth. In the upper part, for instance, the chalcopyrite has in many places been almost completely replaced by chalcocite, with pyrite replaced to a lesser degree. In the lower part of the zone, chalcocite forms thin films along every fracture, however small, but the pyrite is unaffected.

As stated above, the hanging wall limits of the orebody are definitely defined by the tuff contact, but the footwall is purely a commercial limit of workable ore.

METHODS OF PROSPECTING AND EXPLORATION

No systematic exploration was attempted until the property was taken over by Mr. Braden about 1901. Due to the adverse climatic and topographic conditions, he decided that diamond or churn drilling would be impracticable as a method of prospecting, and therefore proceeded with a series of adits, utilizing some of the old workings where they fitted into his scheme. The plan was to drive these adits around the periphery of the crater described, and then crosscut and raise at intervals to determine the lateral and vertical extent of the ore and also the thickness of the lower grade overlying capping.

METHODS OF SAMPLING AND ESTIMATING TONNAGES AND VALUES

Standard channel samples are cut in the sides of the crosscuts, drifts, and raises, each sample representing three meters of working. The assays corresponding to these samples are plotted on assay maps. The ore limits are then laid out on these maps, on the basis of the assays, thus giving the area of commercial ore on each level. The volume between levels (the level interval is ordinarily 50 meters) is then calculated as the frustum of a pyramid. To obtain the grade of this volume, the area on each level is multiplied by its corresponding average assay, thus obtaining an area per cent. These area per cents are then weighted, giving the grade of the ore between the levels. The upper limits of the top level, for instance, are determined by raises and then the volume is calculated from sections, the grade being arrived at by weighting the raise assays with those on the level.

As a method of grade control a grab sample is taken from each active chute in the mine for every 80 tons of ore drawn. Responsible inspectors are employed to observe carefully that no uncommercial ore is pulled from the chutes. They have authority to close immediately any chute showing signs of dilution. These doubtful chutes are promptly sampled on consecutive days, and should the assays obtained show a lower grade than the predetermined cut-off, the chute is definitely abandoned. If these assays show values higher than the required cut-off grade, the chute is then reopened for further draw.

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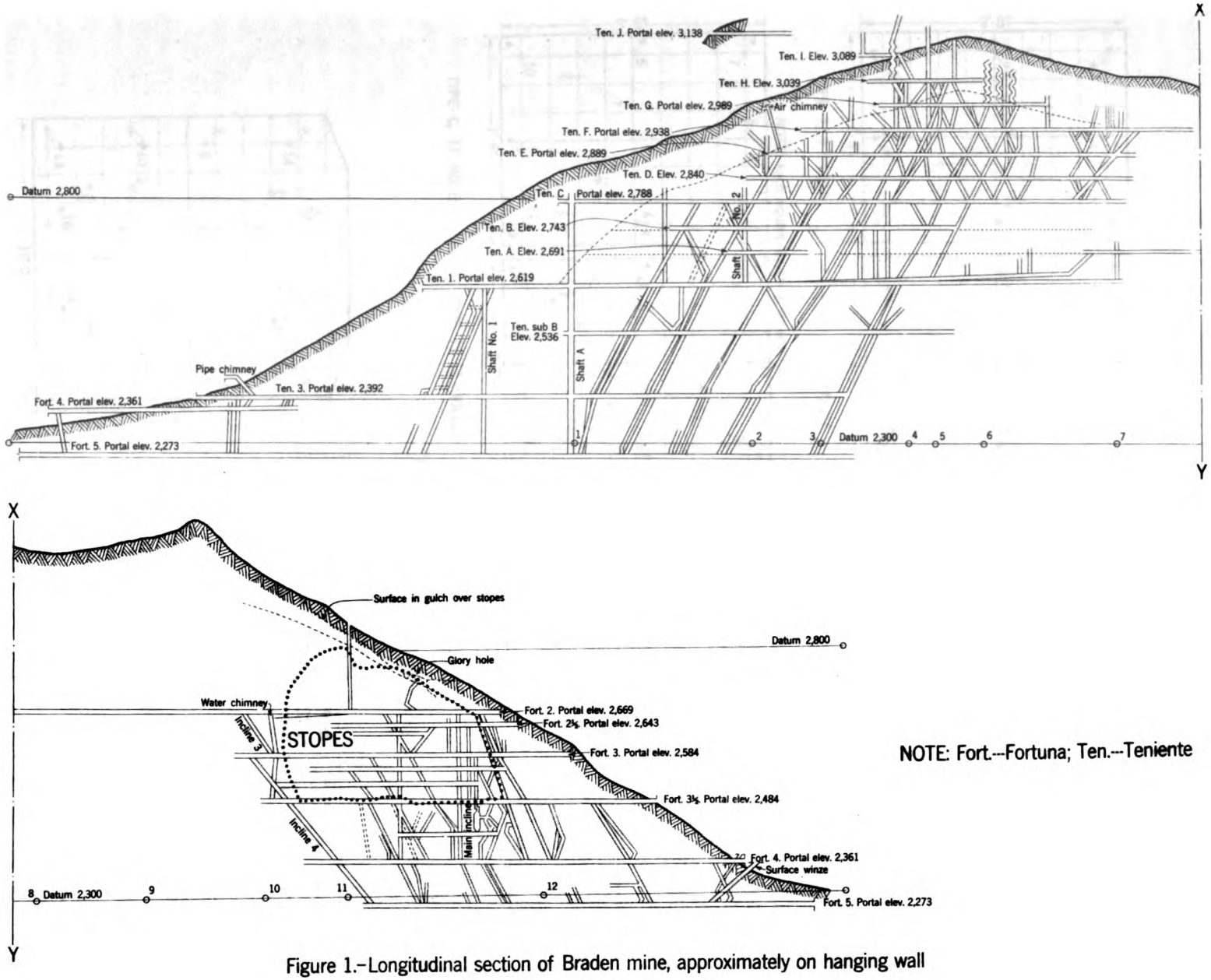

Figure 1.–Longitudinal section of Braden mine, approximately on hanging wall

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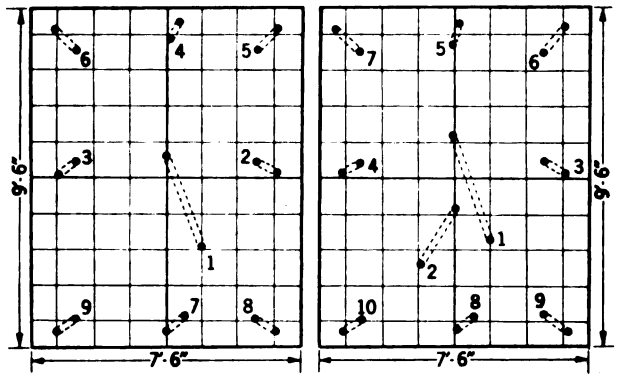


TYPE A, 9 HOLES TYPE B, 10 HOLES

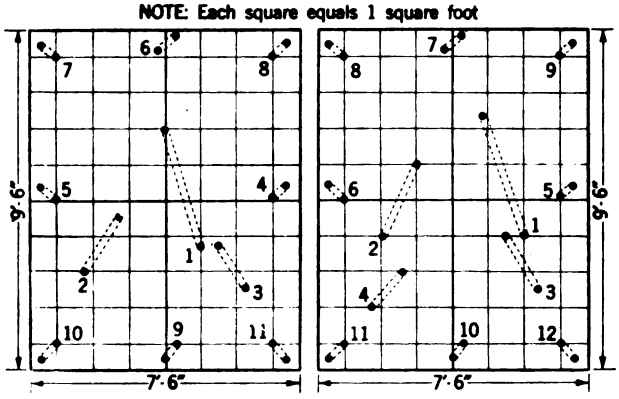
NOTE: Each square equals 1 square foot

Nature of round, type A

Hole No.	Depth, feet	Angle Vert.	Angle Hor.	Powder, sticks	Firing order
1	6.00	25	10	6	1
2	5.25	5	5	5	2
3	5.25	5	5	5	3
4	5.25	5	5	5	4
5	5.25	5	5	5	5
6	5.25	5	5	5	6
7	5.25	5	5	5	7
8	5.25	5	5	5	8
9	5.25	5	5	5	9
Total	48.00			46	

Nature of round, type B

Hole No.	Depth, feet	Angle Vert.	Angle Hor.	Powder, sticks	Firing order
1	6.5	30	10	6	1
2	6.0	15	10	6	2
3	5.5	5	5	5	3
4	5.5	5	5	5	4
5	5.5	5	5	5	5
6	5.5	5	5	5	6
7	5.5	5	5	5	7
8	5.5	5	5	5	8
9	5.5	5	5	5	9
10	5.5	5	5	5	10
Total	56.5			52	



TYPE C, 11 HOLES TYPE D, 12 HOLES

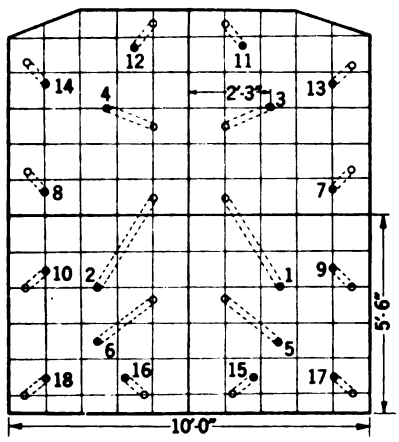
Nature of round, type C

Hole No.	Depth, feet	Angle Vert.	Angle Hor.	Powder, sticks	Firing order
1	6' 6"	30	10	6	1
2	6' 0"	15	10	6	2
3	5' 9"	10	10	6	3
4	5' 6"	5	5	5	4
5	5' 6"	5	5	5	5
6	5' 6"	5	5	5	6
7	5' 6"	5	5	5	7
8	5' 6"	5	5	5	8
9	5' 6"	5	5	5	9
10	5' 6"	5	5	5	10
11	5' 6"	5	5	5	11
Total	62' 3"			58	

Nature of round, type D

Hole No.	Depth, feet	Angle Vert.	Angle Hor.	Powder, sticks	Firing order
1	6' 6"	30	10	6	1
2	6' 0"	20	10	6	2
3	6' 0"	15	10	6	3
4	5' 9"	10	10	6	4
5	5' 6"	5	5	5	5
6	5' 6"	5	5	5	6
7	5' 6"	5	5	5	7
8	5' 6"	5	5	5	8
9	5' 6"	5	5	5	9
10	5' 6"	5	5	5	10
11	5' 6"	5	5	5	11
12	5' 6"	5	5	5	12
Total	68' 3"			64	

Figure 2.—Drift rounds



Nature of round

Hole No.	Depth, feet	Vertical angle	Horizontal angle	Sticks of powder	Order of firing
1	6' 3"	25°	15°	6	1
2	6' 3"	25°	15°	6	2
3	5' 9"	5°	15°	6	3
4	5' 9"	5°	15°	6	4
5	6' 0"	10°	15°	6	5
6	6' 0"	10°	15°	6	6
7	5' 6"	5°	5°	5	7
8	5' 6"	5°	5°	5	8
9	5' 6"	5°	5°	5	9
10	5' 6"	5°	5°	5	10
11	5' 6"	5°	5°	5	11
12	5' 6"	5°	5°	5	12
13	5' 6"	5°	5°	5	13
14	5' 6"	5°	5°	5	14
15	5' 6"	5°	5°	5	15
16	5' 6"	5°	5°	5	16
17	5' 6"	5°	5°	5	17
18	5' 6"	5°	5°	5	18
Total	102' 0"			96	

Figure 3.—Eighteen-hole round, motor haulage drift

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DRAW CONTROL

In mining by caving methods, draw control is one of the most highly important factors in successful operation. At the Braden mine the draw control is in direct charge of the efficiency engineer. Accurate graphs are made showing the draw from each pair of chutes in the mine. For general purposes, these are brought up to date twice a month, but for the closer guidance of the mine operators special large-scale charts, kept in the mine offices, are plotted daily. Chutes that are lagging due chiefly to timber trouble, are more heavily drawn when the timber repairs are completed, the overdrawn chutes being held back in order that the capping may be brought down evenly. Instructions are given to the operating force each day as to the tonnage to be pulled from each chute.

DEVELOPMENT WORK

The Teniente mine has been developed chiefly from adits, shafts, and inclines. Figure 1 shows a longitudinal section. Due to the probable bottom of the orebody being accessible by means of a tunnel, Teniente 5, the ore is dropped from the mining levels down inclined ore passes (raised up through the orebody) to the retram level, Teniente C, where it is motor trammed to other ore passes located in a hard breccia, highly resistant to abrasion, outside of the ore limits, and thence dropped down these latter ore passes to the main haulage level, Teniente 5. Shaft A is used for handling men and supplies from Teniente 5 to Teniente C. Inclines 2 and 4 likewise serve the levels above Teniente C.

The drifts and crosscuts from which the stoping, pillar caving, and undercutting are done are practically all driven in andesite. Standard rounds, determined experimentally, are universally used. The number of holes in these rounds (fig. 2) varies from 9 to 12, depending on the nature of the ground.

In the larger-sized motor-haulage drifts and crosscuts an 18-hole round is used (fig. 3).

All development work is contracted for on a meterage basis, uniform prices having been adopted for each type of drive.

Drilling.- One-man, mounted jackhamers are used in driving practically all horizontal headings, the drilling being done dry. Stopers (hand-rotating type) are employed exclusively in raising.

Drill Steel.- The jackhamers are all operated with ⅞-inch hexagon hollow steel, 1-¼-inch cruciform steel being used with the stopers. The gage of the starters is 2 inches, with a reduction of ½ inch for each succeeding bit. The difference in length between each change is about 12 inches.

Drill Bits.- Double-taper rose bits with angles of 14 and 5° are used. All bits are carefully inspected daily, after sharpening, for gage, sharpness, and form of bit. All steel sharpening is done on contract in underground blacksmith shops equipped with oil forges and pneumatic sharpeners.

Air Pressure.- The air pressure at the compressors is 100 pounds per square inch. At the drills it varies from 80 to 85 pounds.

Blasting.- Primers with fuse and detonators are made up by inserting the cap into the end of the primer cartridge. The primer is always the second stick from the bottom of the hole. All blasting is done with 25 per cent N. G., Ammon-Gelignite. For the standard rounds 1½ by 8 inch cartridges are used. Smaller sticks, ⅞ inch by 8 inches, cut in half, are used for blockholing, bulldozing chutes, and unchoking ore passes. All blasts are detonated with No. 6 lead azid detonators, which have the same strength as standard No. 8 fulminates. These detonators were adopted as standard after a long series of comparative experiments had been carried on. Hot wire lighters are in general use in spitting the fuse. Moist clay cartridges having the same dimensions as the dynamite sticks are used for stemming. These cartridges are made in the mine in a pneumatic machine which forces the clay into waxed paper tubes, thus forming the cartridges.

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DRIFTS AND CROSSCUTS

There are three general types of drifts and crosscuts. (1) For hand tramming, 7 feet 6 inches by 9 feet 6 inches in cross section. (2) For motor haulage, 10 feet by 10 feet in cross section. And (3) for access to control gates and the caving sections for the handling of supplies, 7 feet 6 inches by 9 feet 6 inches in cross section, the same size as for (1) hand tramming; these latter workings being so located when in ore on the lower levels that they can later be utilized as extraction drifts when the mining operations progressively reach the lower horizons.

Hand Tramming Drifts and Crosscuts.- Hand tramming drifts and crosscuts are broken full size to eliminate as far as possible the necessity of widening the drift in order to install the chute sets (fig. 4).

The depth of the round broken averages 1.47 meters (4.82 feet).

The mucking is done by hand into 1-ton cars and on the night shift, the drilling and blasting being done on the day shift. Experiments are being carried on with different types of scrapers and slushers.

Motor-Haulage Drifts and Crosscuts.- Motor-haulage drifts and crosscuts are always timbered wherever the ground shows any signs of sloughing, either from the back or sides. As has been noted, a standard 18-hole round is used, shown in Figure 3, the drilling being done with one-man, mounted jackhamers using ⅞-inch hexagon hollow steel.

The mucking here is also done by hand into 5-ton cars. Some experimentation is being carried on with mechanical loaders.

Drifts For Access to Control Gates, Handling Supplies.- In so far as their driving is concerned, drifts for access to control gates can be classified with those for hand tramming.

RAISES

Ore Passes.- At Braden it has been found that the ore transfer raises or ore passes, as they are locally known, operate best when driven on an angle of about 60° with the vertical.

These ore passes are classified as: (1) Main ore passes, those located in the hanging-wall breccia and running from Teniente 5 to Tiente C; and (2) retram ore passes, those running from the retram level to the producing level, generally through the ore-body. The accompanying sketches (fig. 5) show each type.

Main Ore Passes.- Main ore passes and their auxiliaries are always raised in the hanging wall, in a tough breccia, which, as has been noted before, is highly resistant to the abrasion of the falling ore. These raises are run 2 by 2 meters (6.56 by 6.56 feet) in cross section. The round consists of 16 holes loaded with approximately 94 sticks of 25 per cent N.G., Ammon-Gelignite, giving a break of 1.20 to 1.50 meters (4.25 to 4.92 feet). Stoper machines are used, two miners working together and sharing in the contract, which is let on a meterage basis.

The general method of raising one of these ore passes is as follows:

When the ore pass has attained a slant height of 5 meters (16.42 feet) above the level, the miners drive two holes in the floor, normal to the floor, 1.5 meters (4.92 feet) down from the face, into which they insert two pieces of 1-inch round iron, leaving 2 feet projecting out in the ore pass. Against these supports is placed a 2 by 12 inch by 5 foot board which serves as a platform for the miners and their machines while drilling the round. Attached to one of the above pieces of projecting iron is a 1½-inch diameter hemp rope, with the help of which the miners climb up the floor of the ore pass to the working face. As the raise advances, the platform with its iron supports is moved up, being kept at all times

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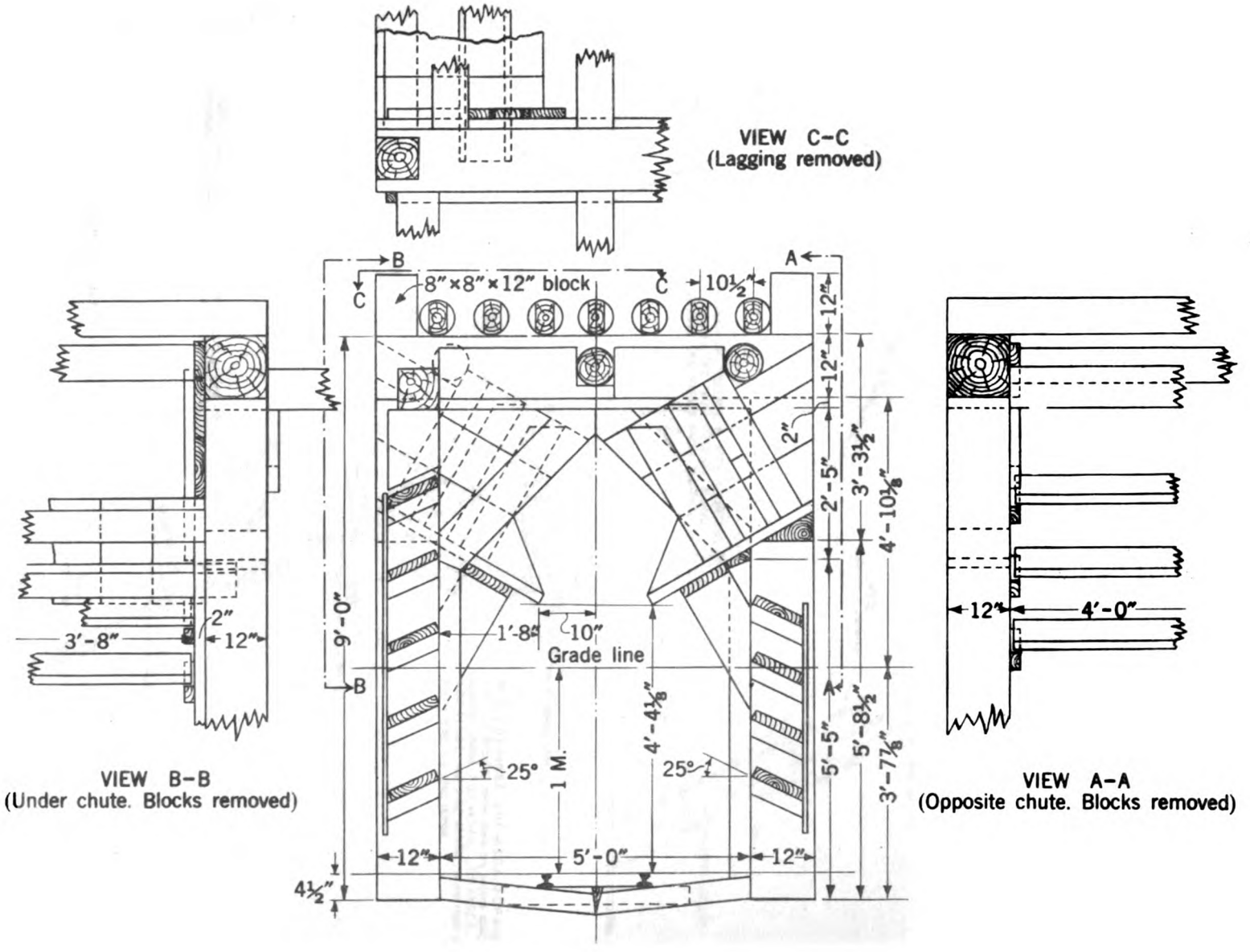

Figure 4.— Standard chute set

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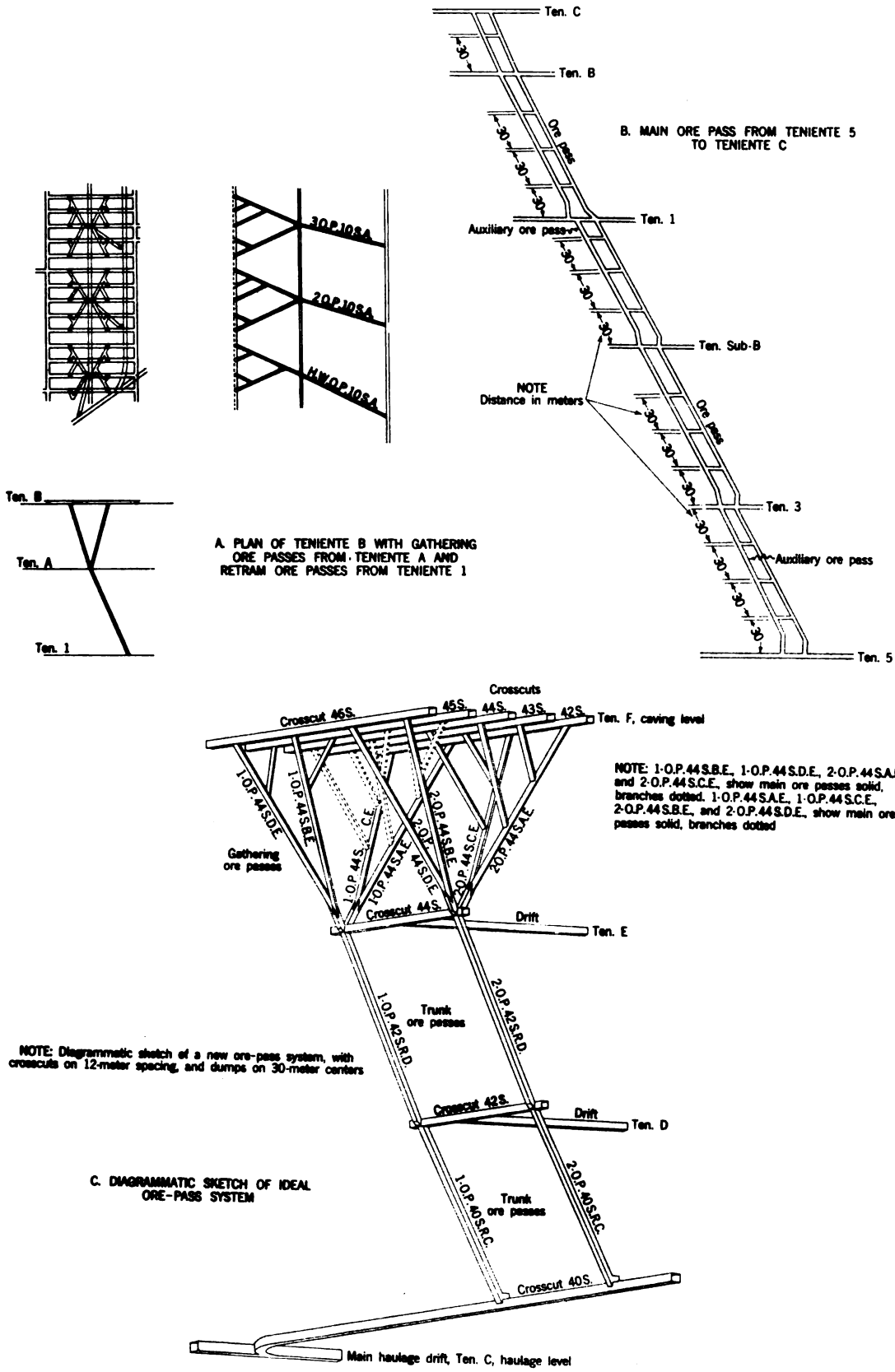
Ten. C
Ten. B
Ore pass
B. MAIN ORE PASS FROM TENIENTE 5 TO TENIENTE C
Ten. 1
Auxiliary ore pass
Ten. Sub-B
NOTE
Distance in meters
Ore pass
Ten. 3
Auxiliary ore pass
Ten. 5
30 O.P. 10 S.A.
2 O.P. 10 S.A.
HW O.P. 10 S.A.
Ten. B
Ten. A
Ten. 1
A. PLAN OF TENIENTE B WITH GATHERING ORE PASSES FROM TENIENTE A AND RETRAM ORE PASSES FROM TENIENTE 1
Crosscuts
Crosscut 46S.
45S.
44S.
43S.
42S.
Ten. F, caving level
NOTE: 1-O.P. 44 S.B.E., 1-O.P. 44 S.D.E., 2-O.P. 44 S.A.E., and 2-O.P. 44 S.C.E., show main ore passes solid, branches dotted. 1-O.P. 44 S.A.E., 1-O.P. 44 S.C.E., 2-O.P. 44 S.B.E., and 2-O.P. 44 S.D.E., show main ore passes solid, branches dotted
1-O.P. 44 S.D.E.
1-O.P. 44 S.B.E.
1-O.P. 44 S.C.E.
2-O.P. 44 S.B.E.
2-O.P. 44 S.C.E.
2-O.P. 44 S.A.E.
Gathering ore passes
1-O.P. 44 S.A.E.
44 S.D.E.
Crosscut 44S.
Drift
Ten. E
Trunk ore passes
1-O.P. 42 S.R.D.
2-O.P. 42 S.R.D.
NOTE: Diagrammatic sketch of a new ore-pass system, with crosscuts on 12-meter spacing, and dumps on 30-meter centers
Crosscut 42S.
Drift
Ten. D
C. DIAGRAMMATIC SKETCH OF IDEAL ORE-PASS SYSTEM
1-O.P. 40 S.R.C.
Trunk ore passes
2-O.P. 40 S.R.C.
Crosscut 40S.
Main haulage drift, Ten. C, haulage level

Figure 5.—Types of ore passes

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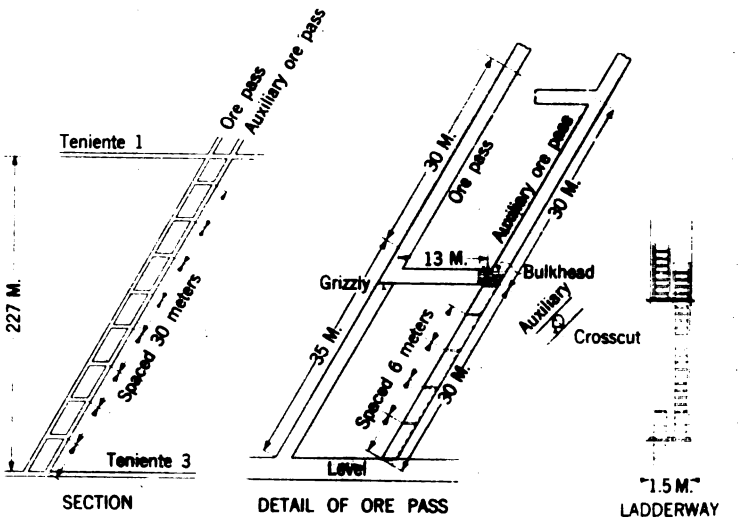


Figure 6.—Method of driving ore pass

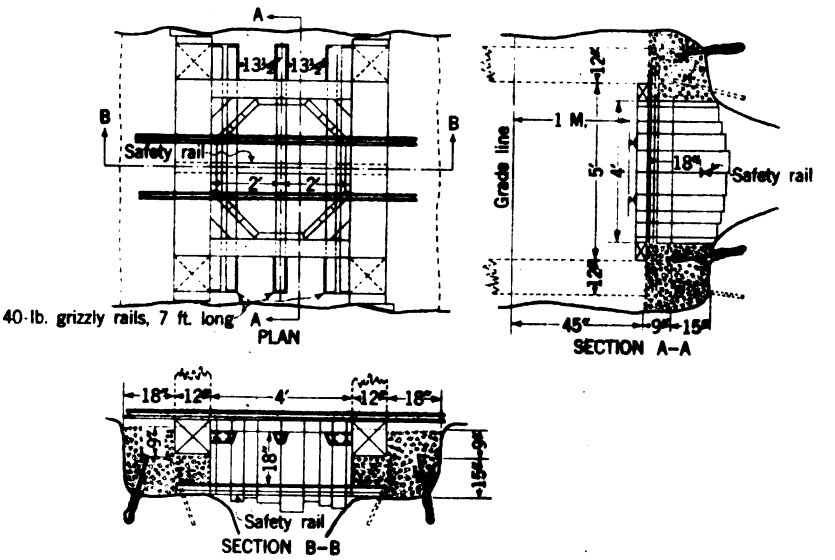
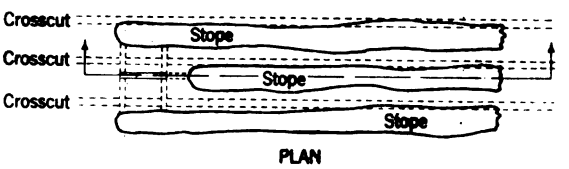


Figure 7.—Standard grizzly



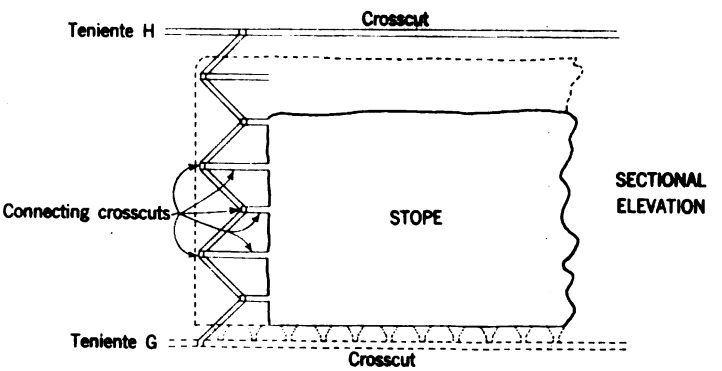


Figure 8.—Turret raise

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within 1.5 meters (4.92 feet) of the face. The compressed air to supply the machines is carried up the raise in a 2-inch air line, which is progressively extended to within 20 feet of the face as the raise advances.

When the ore pass and auxiliary have both been raised for a distance of 35 meters (114.75 feet) they are temporarily stopped and a connecting crosscut is driven (fig. 6). When this connection is made, a grizzly is set in the ore pass, level with the floor of the crosscut. In the auxiliary a bulkhead is placed 2 feet above the floor of the connecting crosscut and extended into it, enabling the miners to get from the auxiliary, underneath the bulkhead, into the crosscut. This bulkhead further serves as a floor on which the broken material from the blasts in the auxiliary is caught. This muck is trammed through the connecting crosscut and dumped into the ore-pass grizzly. A ladder way is placed in the auxiliary from the level to the bottom of the bulkhead, affording a convenient entrance for men and supplies. These connecting crosscuts are driven at intervals of 30 meters (98.4 feet) and the same procedure is continued throughout the raising of the ore pass and the auxiliary. The ultimate use of the auxiliary in addition to the safety feature during the driving as outlined above, is to provide a secure means of access for inspection of the ore pass and to enable chute blasters to place charges in the ore pass when it becomes choked due to an accumulation of fines or too high a moisture content in the ore. As soon as an ore pass has been driven up a few meters, a "Cousin Jack" chute is put in on the haulage level to eliminate any shoveling. The ore pass is connected from the level below to the one above, and a grizzly is placed before continuing with the raise, thus eliminating any handling of the muck on the intermediate levels.

Retram Ore Passes.- Retram ore passes are driven in the same manner as those on the lower levels and with the same cross section. However, due to the shorter lifts the upper level raises are put up without the accompanying auxiliaries and the small connecting crosscuts. As the retram ore passes are generally located in andesite, a 5-foot round can usually be broken with nine holes, each loaded with a charge of 5 sticks of 25 per cent N.G., Ammon-Gelignite.

Winzes to connect with these ore passes are sunk from the level above to a depth of 2.5 meters (8.2 feet). The winzes are driven as small as possible and enlarged, after holing through, to such size as will permit the permanent grizzly with its concrete reinforcement and safety rail to be securely and accurately placed (fig. 7).

The object of driving the winze of extremely small size is to avoid a large overbreak in the drift or crosscut, which would necessitate a very large grizzly sill and introduce the possibility of excessive maintenance cost due to the grizzly sill taking weight. Under the present method of sinking very small winzes and later enlarging them, after the connection has been made the grizzly sills are set on solid ground, and only in exceptional cases are failures experienced.

Exploration Raises.- Exploration raises are put up on a 45° slope with a cross section of 1.50 by 1.50 meters (4.92 by 4.92 feet). Wherever practicable they are planned to be used later as observation raises for noting any caving action of the block undercutting; and also when necessary for ventilation, they are broken through to the surface or connected with the level above.

Pilot Raises.- Pilot raises are used in making shaft connections. The shafts are located in a hard breccia, and the method of raising, in a large-sized shaft, is as follows:

The pilot raise is zigzagged up on a 50° slope, being kept within the shaft section. After breaking through to the top level the raise is enlarged to the full shaft size from the top down, the shaft timber being carried down with the enlargement, and the spoil dropping down through the pilot raise to the level below.

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SHAFTS

Shafts 1 and 2 were sunk and raised to provide for handling men and supplies, between Teniente 5 and Teniente C, during the development period of the Teniente mine.

Shaft A was completed in 1920 to take care of the increased traffic for mine production. This shaft has two hoisting compartments 9 by 7 feet 1 inch in size each equipped with a 50-man, double-deck cage and a manway of 9 by 6 feet, up which are carried air, water, and power lines and a ladder way. The shaft is timbered with 10 by 10 inch Oregon pine sets, spaced on 6½-foot centers. As a protection against fire this shaft is equipped with an automatic sprinkler system. The hoisting equipment consists of a double drum hoist, operating in balance, driven by a 160-hp. A.C. motor. The drums are 7 feet in diameter by 4 feet 6 inches long and are grooved for 1⅛-inch cable. This equipment is provided with both overwind and overspeed safety devices.

MINING METHODS

In the early days the higher-grade portions of the outcrops were first worked intermittently, all ore being hand sorted and sacked. Later, the high grade having become depleted, a tunnel was driven in Fortuna 2, and a raise put up to the surface. This showed ore running about 4 per cent copper, but as the ore was well disseminated, it was impossible to handle it at a profit without concentration. Mr. Braden, becoming interested at about this time, put up a 250-ton mill and ore was obtained first by glory-hole methods and then, as this gradually became impracticable, by shrinkage stoping.

The original Fortuna stopes were planned to run normal to the long axis of the orebody with a width of 7 meters (22.95 feet) having a 10-meter (32.8 feet) pillar between them. In the center of these pillars starting alternately from the footwall and hanging-wall boundaries the stope entrance chimneys were raised on a 45° slope, zigzagging, in the plane of the pillar, from the footwall toward the hanging wall or vice versa. These chimneys gave access for men, supplies, and the air line to the shrinkage stope on both sides by means of small crosscuts spaced 7 meters (22.95 feet) vertically. The height of these shrinkage stopes ran up to approximately 100 meters (328 feet). It was found, however, in subsequent pillar caving operations, that it was difficult to break such thick pillars, due to their tendency to arch longitudinally so as to necessitate raising small stopes in the footwall and hanging-wall extremities of the pillar. Because of this unfavorable condition, when stoping was started in the Teniente mine the thickness of the pillars was decreased to 5 meters (16.4 feet), leaving the width of the stope at 7 meters (22.95 feet). This required a change in the stope entrance chimney. A "turret raise" was then substituted. This raise was driven on a 45° slope with 8-meter (26.23 feet) legs, one raise being located at each end of a set of three stopes and serving them by small connecting crosscuts (fig. 8).

The successful caving of these pillars still presented difficulties, and a further reduction in their width was made to 3 meters (9.84 feet), reducing the stope width to 5 meters (16.4 feet).

Cribbed manways were then adopted for stope entrances, these being raised in solid rock at each end of the stope and abutting the slope. These manways are equipped with ladders and are used as an entrance for men, supplies, and a 2-inch air line. At the foot of the manway a tugger hoist is located, by means of which steel and other supplies are hoisted to the top of the stope (fig. 9).

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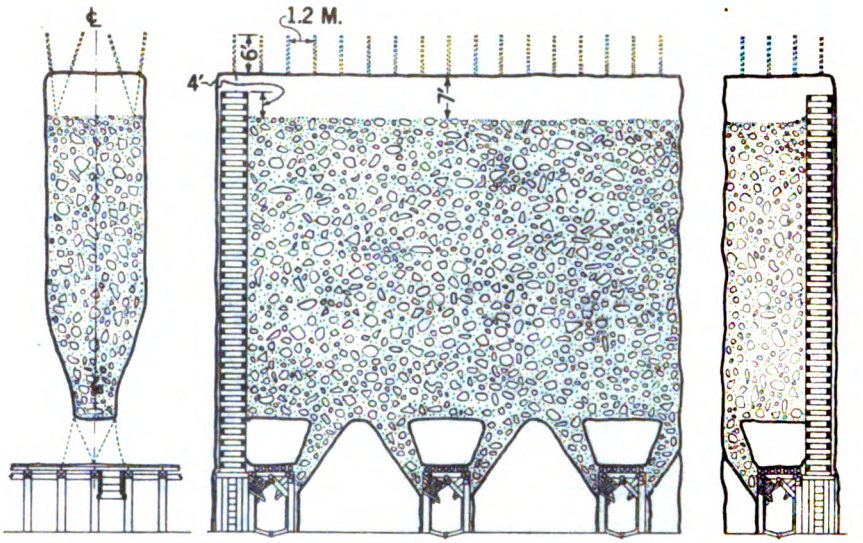

Figure 9.—Spacing of holes in stope

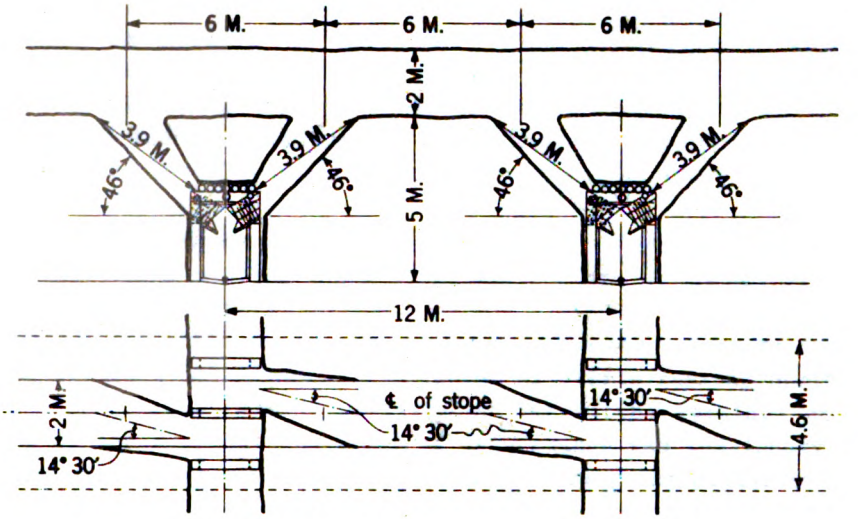

Figure 10.—Stope chute raises. Stope raised on new system normal to drift

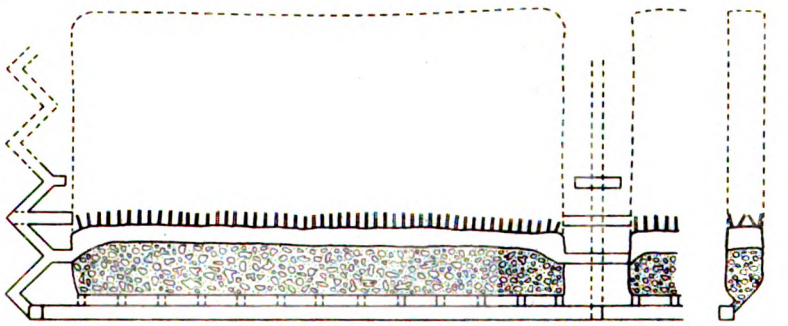

Figure 11.—Stope raised on old system parallel to crosscut

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STOPING

The drifts under the stope having been driven, the first step is to put up the stope chute raises from these drifts (see fig. 10). These are located on 6.1 meters (20 feet) centers and are driven up 3.90 meters (12.8 feet) on a 46° slope at an angle of 75°30' from the center line of the drift to the elevation of the stope sublevel. After driving about 1 meter (3.28 feet) of the stope chute raise (one round), the timber for the chute is stood and the chute box installed. The tops of the stope chute raises are then connected by a drift, 7 feet high and 6 feet wide, from the hanging-wall to the footwall limits of the stope. The chutes are then belled out to insure good drawing, and the stope is started by widening out the drift and raising from it.

Approximately 40 per cent of the tonnage of the ore broken out of the solid is drawn for production, the remaining 60 per cent being left in the stope to support the pillars and allow the miners to drill the back for the next round.

The stope crew drill and blast 50 meters (164 feet) along the stope during each shift. The number of miners required is roughly calculated at 10 per stope, thus giving each pair about 10 meters (32.8 feet) of back to drill. The holes are spaced along the length of the stope from 1 to 1.25 meters (3.28 to 4.1 feet) apart; they are drilled in sets of four consisting of two cut holes and two side holes, the cut holes being given an inclination of about 20° from the vertical and the side holes being sloped slightly toward the walls. The average depth of hole is 6 feet and the average number drilled per machine shift varies from 15 to 20. An average of 70 tons per machine shift is broken, of which 28 tons can be drawn as shrinkage.

All the men in the stope, including the stope boss, stopemen, mechanics, and tool nippers, work on contract, the monthly basis of payment being the amount of broken tonnage as calculated by the engineers.

With the changes in the sizes of the stopes and pillars the system of crosscuts driven from hanging wall to footwall was changed to a system of drifts running north and south or perpendicular to the crosscuts. The orientation of the stopes remained as before, thus bringing them normal to the drifts. The basis of this change was the theory that the compressive stresses set up by the caving of the pillars, during the pillar caving operation, crushed the pillar stubs, in stopes raised parallel with the crosscuts, thereby throwing the entire overhead weight on the crosscut timbers. With the stopes normal to the drifts it was reasoned that the lugs between the drifts would not break up as readily and would carry more of the weight of the broken mass, thus relieving the pressure on the drift sets and decreasing the timber costs. Actual practice has borne out this assumption, by a reduction in the cost of timber maintenance. A further advantage of having the stopes run normal to the drifts is the greater amount of ground available at all times for extraction. With the old system (fig. 11) it was necessary to cave from 60 to 90 meters (197 feet to 295 feet) of the crosscut before draw operations could be started. In heavy ground often as much as half of this available drawing area was down due to timber failures in the crosscut. This greatly reduced production from the section under draw. With drifts, and stopes normal to the azimuth of the drifts, it is seldom necessary to have over 30 chutes of 45 meters (147.6 feet) undercut at one time. Caving from hanging wall to footwall, the various drifts give more points of access for drawing. If one or more drifts have timber trouble, seldom, if ever, is the work of more than two loaders interfered with, whereas under the old system from six to ten loaders might be unable to draw in the section affected.

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PILLAR CAVING

The work of pillar caving, consisting of breaking up the solid pillar between the stopes, is commenced by putting up the chute raises 3.50 meters (11.48 feet) into the pillar. The tops of these raises are then connected by a 2 by 2 meter (6.56 by 6.56 foot) drive for the entire length of the pillar. Over a 6-meter (19.67-foot) length the top and sides of this drive are enlarged, and holes are then drilled to within breaking distance of the adjoining stopes. These holes are blasted, with the resultant breaking up of the base of the pillar. This procedure is continued for the full length of the stopes until the entire pillar base is weakened. The subsequent drawing of the chutes, underneath, furthers the complete disintegration of the pillar. This process is graphically shown in the accompanying sketches. (Fig. 12.)

BLOCK UNDERCUTTING

The block undercutting method of mining is the gradual development of a system evolved at Braden from the experience gained in the caving of pillars between shrinkage stopes. The method can be defined as the process of caving virgin ground adjacent to country that has previously been weakened or caved by a minimum amount of stoping and pillar caving.

In shrinkage stoping, as has been noted before, it is necessary to raise stope chutes and connect their tops with a small crosscut. With the present caving practice this raising and driving is eliminated, resulting in a corresponding saving in the preparation work. This excessive preparation work and the high cost of breaking ore in stoping were the main factors in bringing about the change in caving methods, the new system being obviously more economical. The cost of ore broken in shrinkage stoping has averaged 43.37 cents per ton while block undercutting costs have averaged 20.5 cents per ton, a direct saving of 22.87 cents. With this marked reduction in cost by block undercutting as compared with stoping, the present practice is toward raising a minimum number of shrinkage stopes, mainly for boundary cut-offs and such preliminary stope weakening as is necessary on a new level. Of the present delivered tonnage, over 85 per cent is derived from block undercutting.

The undercutting is commenced by taking a section of a drift 6 meters (19.68 feet) long and drilling a 6-foot round above the top lagging and through the drift chutes, as shown in Figure 13 at A. The ore broken in this first round is drawn out through the drift chutes, leaving a small open stope approximately 7 feet high above the drift timbers. Access to this stope, after the first blast, is through the end chute next to the virgin country in the section of drift blasted. Preliminary to this initial blast, an entrance chimney has been completed from the adjacent drift to the center of the pillar, as shown in the sketches. This entrance is driven with a 1.50 by 1.50 meter (4.92 by 4.92 feet) section on a 48° slope with an angle of 8° 30' from the perpendicular to the drift to meet a similar chimney driven from the last chute in the undercut stope, as shown. Before making the second widening blast in the undercut stope, the entrance chimney is connected, thus providing a safe entrance for men for subsequent shots in the undercut stope. A second and third widening blast are made, as shown in Figure 13, B and C. Upon making the third blast, the ground usually shows signs of weakening, and it is necessary to stull up the back in order to assure safety for the miners in drilling and loading the final blast which breaks to the adjacent caved country, as shown in Figure 13 D. This operation is carried on in progressive slices in successive drifts until sufficient country is weakened to produce continuous caving action. After an undercut section has been completed in a drift, 300 tons of ore per chute are drawn to allow sufficient room for further caving action. These chutes are then sealed and remain closed until the next progressive slice has been undercut. The reason for

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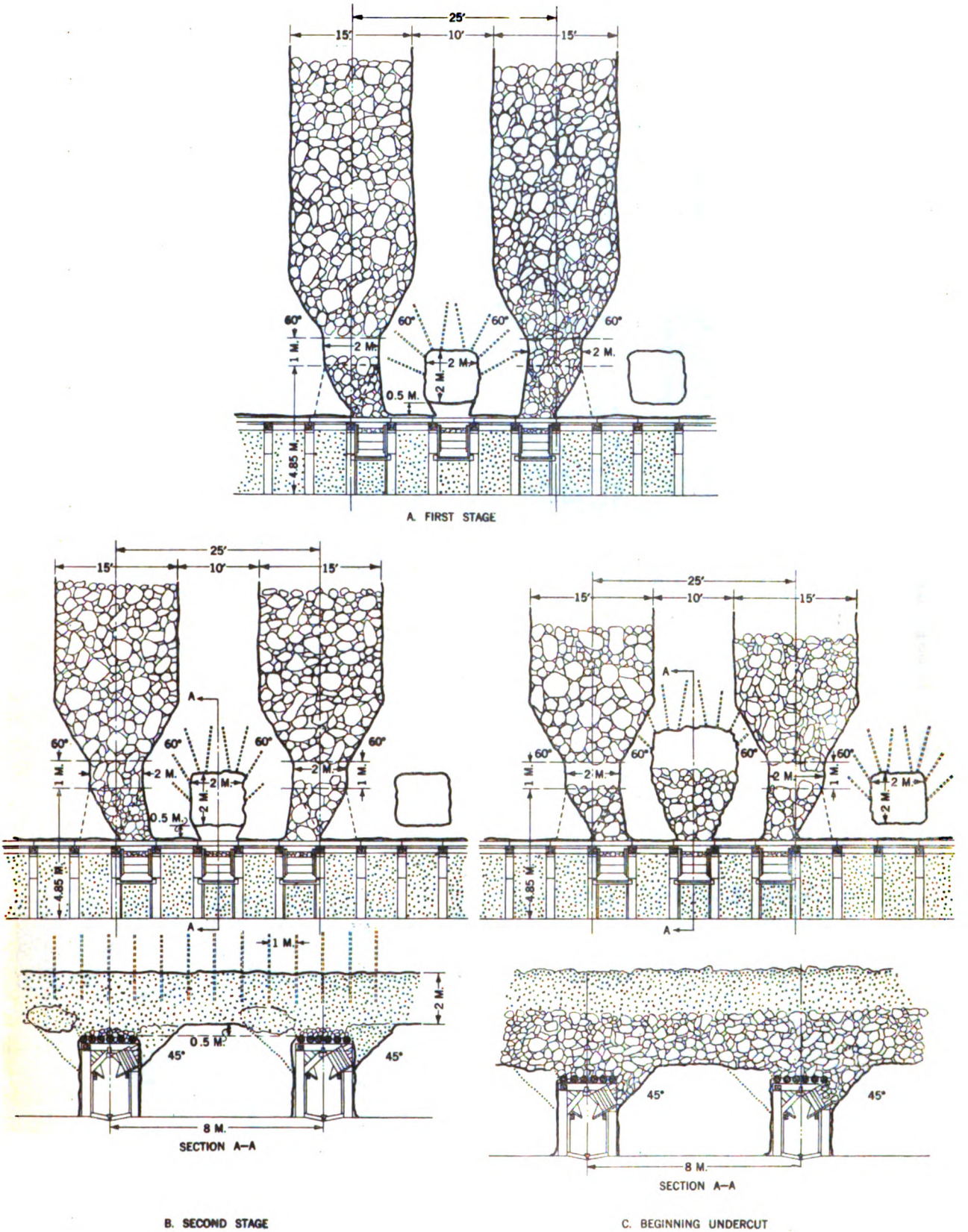

Figure 12.— Pillar caving drifts replacing crosscuts

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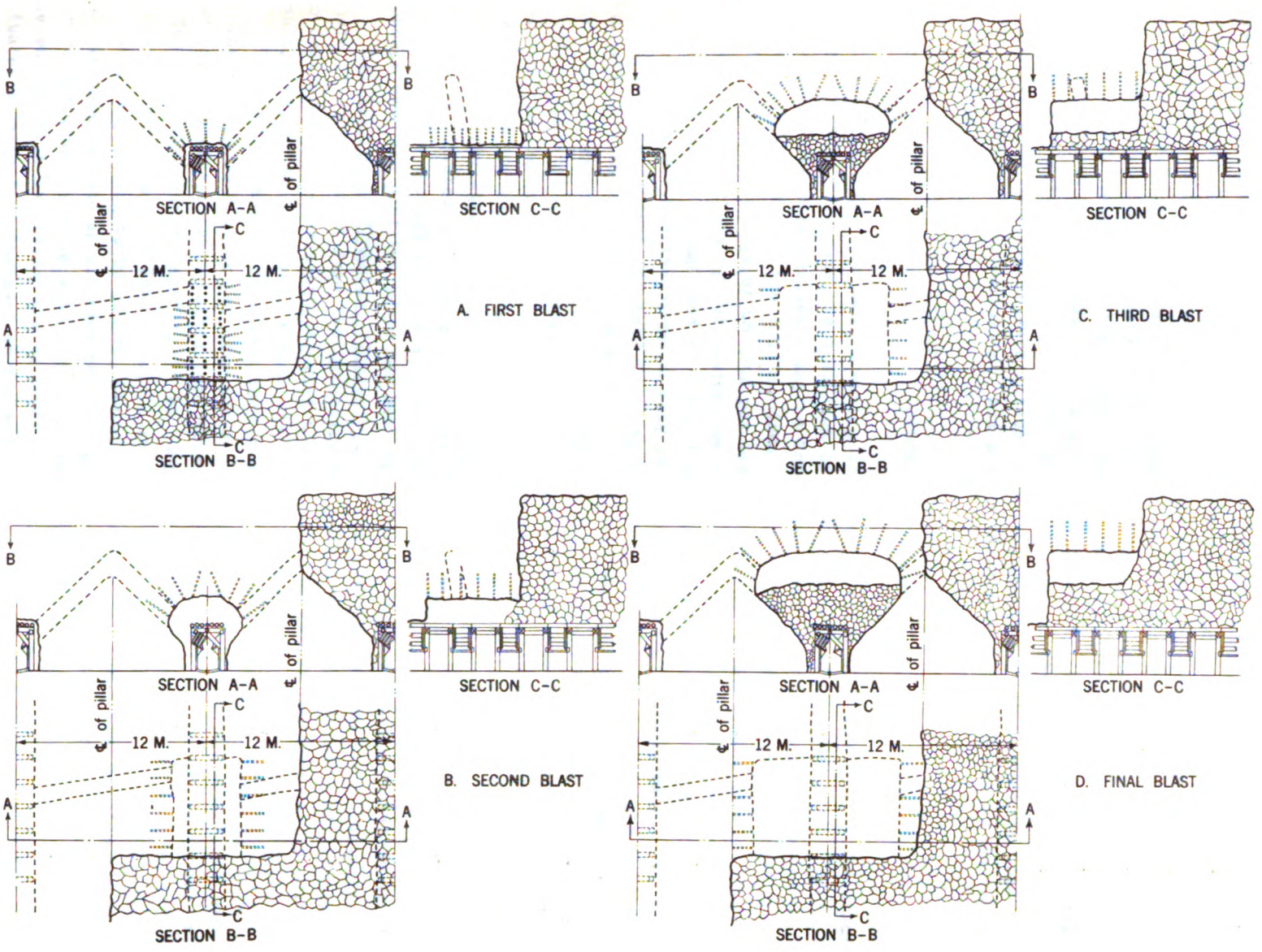

Figure 13.—Undercutting practice

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this procedure is to keep the miners from working against an open stope. That portion of the entrance chimney, leading into the last undercut stope, is used as an observation entrance to note the caving action in the preceding slice. Where a pillar is unavoidably left between this undercut slice and the preceding slice, and where the back is firm enough to allow stulling, a passageway is opened up to allow redrilling the unbroken pillar.

An undercut crew consists of four miners who are paid on contract on a lineal meterage basis of drift undercut. Stoping machines are used exclusively for all drilling.

Formerly it was considered at Braden that 50-meter (164 foot) lifts were the maximum that could be caved successfully. Due to a study made on the action of caving ore, however, 100-meter (328 foot) lifts were attempted and successfully caved, with no loss of ore and with no excessive waste dilution. All present development is laid out on the basis of 100-meter lifts.

UNDERGROUND HAULAGE

Hand Tramming.- All hand tramming is done by means of 1-ton cars. The trammers are placed so as to give each one a minimum of 10 chutes to pull in order that a daily tonnage, specified by the draw engineer, may be drawn from each chute. Dumps are spaced on 30-meter (98.4-foot) centers in the drifts, thus giving each trammer a maximum tram of 15 meters (49.2 feet). An average of 55 tons per trammer is produced per 8-hour shift. Chute blasters aid drawing operations by bulldozing the chutes and generally helping the trammers. Usually one chute blaster is employed to every 3.5 to 4 trammer, the ratio depending on the kind of ore being drawn. Twenty-one-pound baby jackhamers are used in bulldozing.

Since the payment for hand tramming in caving sections is made on a contract basis it was necessary to have an accurate count of cars trammed by each man. Formerly this record was kept by car-checkers. This system proved unsatisfactory and an automatic car counter, operated by an arrangement of springs and cams, was designed and installed on each car used in the caving sections. As the cars are of 1-ton capacity, it is necessary for the trammers to load this weight into the car and actually dump it before the automatic device will register. The markers on each car are read at the beginning and end of each shift. The number of cars paid for is based on the difference in the readings of the counters. The success of this automatic counter is borne out by the fact that the produced tonnage over the past two years has been equal to the weighed tonnage delivered to the mill.

Electric Motor Tramming. (Teniente C).- The ore is retrammed on Teniente C level, where it is collected from the gathering ore passes which serve the producing levels, and is delivered to the main hanging-wall ore passes. The latter run down to the main electric haulage way on No. 5 level, whence the ore is delivered to the coarse-ore bins (fig. 1).

The retram haulage equipment consists of 10-ton locomotives, driven by D. C. motors, having a speed of 8 miles per hour and a capacity load of fifteen 5-ton ore cars per train. The cars are of the gable-bottom type with an overall length of 11 feet 7½ inches, width of 5 feet 2 inches and a capacity of 95 cubic feet. They are provided with either hand or automatic air dumpers. Each electric locomotive is equipped with an air compressor, type CP 28-A, compressing to 90 pounds, having a capacity of 25 cubic feet of free air per minute and discharging into an air receiver having a capacity of 7.5 cubic feet. From this receiver air connections are made with the train through standard railroad air-line equipment. On one end of each car is an air cylinder, controlled by a 3-way valve, the piston of which actuates a ratchet arrangement on the end of the piston arm engaging with a half gear. This half gear is connected to the door lever arms and controls them. By standing at the side of the train and operating the 3-way valve a man can open and close the side of the cars over the main ore pass dumps, thus eliminating the dangerous practice of climbing in between

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the cars and manually operating the dumping device. The time consumed in dumping the trains is also lessened, having now been decreased to one minute. Roller bearings are standard equipment for this section. The track has a gage of 30 inches, 40-pound rail being used. All switches throughout the level are governed by semiautomatic signals. The standard loading gate used on the level allows a 15-car train to be loaded in four minutes. The dumps on this level are 17 feet long and 14 feet wide. The grizzlies are made with 60-pound rail spaced to give a 14-inch opening. Train movement throughout the level is controlled by a train dispatcher. A special telephone system, installed exclusively for this service, has connections at each crossing, ore pass, and dump. By means of this system, train orders are communicated to the train crews.

Operations are conducted on two 8-hour shifts with four 15-car trains, allowing a 4-hour interval between shifts in which to take care of such items as track maintenance and cleaning up. Payment of motor crews is by contract based on monthly tonnage hauled. The handling of the ore through the ore passes above Teniente C is controlled by gate tenders working on the intermediate levels and under the supervision of the retram foreman. Figure 14 shows the type of control gate used on the intermediate levels.

Electric Motor Tramming. (Teniente 5).- On Teniente 5 level the ore is transported from the main ore passes, coming down from Teniente C, to the mill coarse-ore bins located on the surface. The gage of the track in this haulage way is 30 inches but 60-pound rails are used to take care of the heavier traffic. The grade is 0.5 per cent in favor of the loaded ore trains. Twenty-five-ton electric locomotives, operating on 550-volt direct current, having a maximum speed of 7 miles per hour with a drawbar pull of 9,000 pounds at this speed, and equipped with air and hand brake control, are used exclusively in this section of the mine. The maximum ore haul is 3,886 yards. Trains make a complete round trip, including weighing and dumping, from the main ore passes to the mill bins in 30 minutes The ore cars used are of the gable-bottom type, having a capacity of 23.8 tons. The trains are equipped with automatic air control. They consist of 12 cars and operate singly on the return trip from the mill bins to the ore passes and in tandum on the outgoing run.

The train crew consists of a motorman and two brakeman, who assist in loading at the ore passes and in the dumping operation at the bins. All switches are governed by electric semaphore signals and train movements are controlled through a central dispatching office, where each train crew reports to receive train orders.

Supply and passenger trains are also operated at regular intervals over this road. Steel passenger cars, of the Broadway type, are used for passenger service.

HOISTING

Hoisting is confined to the handling of men and supplies, all vertical ore handling being done by gravity through the ore passes.

WAGE, CONTRACT, AND BONUS SYSTEMS EMPLOYED

In 1928 seventy per cent of all underground mine labor was paid on a contract basis.

A standard set of contract prices is in effect throughout the mine, covering all such work as drifting, raising, winzing, stoping, undercutting, tramming, steel sharpening, and timbering, both for standing new sets and for repair work.

All men working on contract are guaranteed day's pay earned at the base rate of the occupation involved.

A bonus is paid on rush work, such as driving drifts or tunnels, where speed is an important factor. This is arranged on a meterage basis, an additional payment being made

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for every meter driven over and above the predetermined monthly average for the particular type of ground in which the work is being done.

VENTILATION

No mechanical ventilation has ever been found necessary. Circulation is always good due to the system of ore passes, with their connecting auxiliaries, various shafts and inclines, and adits and raises broken through to the surface.

The air is always cool and the quantity good.

DRAINAGE

All drainage is by gravity to the various portals of the mine, no pumping being required

RESCUE WORK

Classes in mine rescue work are conducted by the safety engineer. Periodic drills, giving instruction as recommended by the United States Bureau of Mines, are given to all bosses.

These drills cover the use of oxygen helmets and all other rescue apparatus.

SAFETY METHODS
(First-Aid Organization and Training)

Accident prevention and the safety of workmen have always been considered as major problems.

First-aid classes are conducted the year round by the safety engineer for the bosses and workmen.

A safety inspector makes daily mine inspections and recommendations with regard to any hazards or dangerous practices observed.

The use of hard boiled hats is compulsory throughout the mine. As a protection against hand wounds, armoured gloves are furnished to all trammers. For work in ore passes and other hazardous locations, the workmen are provided with safety belts. Chute blasters and miners are supplied with canvas bags in which to carry their powder.

A bonus system, based on the severity rate of accidents, has been devised, whereby the bosses have an incentive to enforce all safety rules and measures in vogue in the mine.

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SUMMARY OF COSTS

Teniente mine — Year, 1928.
5,174,017 tons delivered
Mining method, block undercutting

	Labor[1]	Air drills and steel[2]	Power	Explosives	Timber	Other supplies	General expense	Total
Development, cents per ton	1.01	0.39	-	0.64	0.33	0.20	2.06	4.63
Mining (undercutting) ...do.	7.77	0.89	-	1.52	4.15	0.63	4.14	19.10
Hand tramming .....do.	3.93	-	-	-	-	0.27	-	4.20
Ore handling[3] .....do.	2.14	-	0.04	-	0.28	0.77	-	3.23
Transportation (E.R.R.)[4] do.	1.00	-	0.10	-	-	0.56	-	1.66
General expense .....do.	-	-	-	-	-	-	15.55	15.55
Total cost.....do.	15.85	1.28	0.14	2.16	4.76	2.43	21.75	48.37

1 Labor includes supervision.
2 Includes power for compressors, maintenance of drilling machines, new drill steel, and cost of drill sharpening.
3 Includes retramming on Teniente C and passing ore from the producing levels to the main haulage level.
4 Includes only electric haulage from main ore pass gates to mill bins.
5 Of total general expense, approximately one-half covers charges applicable to the mine proper, such as sampling, fire protection, safety measures, efficiency, engineering, water, service handling supplies, office and hospital expense, and sundries. The balance consists of outside charges, such as Rancagua and New York office expense, taxes, insurance, welfare, legal expense, etc.

COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

Year	Dry tons mined	Tons per man	
		Direct operation underground	Total pay roll, including surface and shops
1927	5,288,310	13.2	9.2
1928	5,174,017	12.2	8.7

Power, kilowatt-hours per ton shipped

Item	Kilowatt-hours per ton
Retramming (Teniente C)	0.15
Electric railroad	0.34
Air service	1.85
Steel sharpening	0.01
Mine, general (sampling, hoisting, water service, lighting, heating)	0.56
Total	2.91

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Explosives
(Ammon-Gelignite 25 per cent N.G.)

	Tons broken per pound of powder	Pounds of powder per ton broken
Primary Blasting:		
Development	0.83	1.205
Undercutting (including stoping)	28.30	0.035
Secondary Blasting:		
Chute blasting on drawing levels	35.7	[1]0.028
Ore-pass blasting	98.6	[2]0.010
	Tons delivered to mill per pound of powder	**Pounds of powder per ton delivered to mill**
Total explosives	8.62	0.116

1 - Bulldozing chutes. 2 - Unchoking ore passes.

Timber

Timber, bd. ft. per ton, 1.08 ..... Based on total tonnage delivered to mill and includes timber used in all mine operations.

DETAIL OF COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

Development

	Drifting	Cross cutting	Raising	Total for all development
Specifications:				
Size of excavation	7'6" x 9'6"	7'6" x 9'6"	6'6" x 6'6"	-
Timbered, or not	Yes	Yes	No	-
Physical properties of rock	Medium	Medium	Medium	-
Labor (man-hours per foot):				
Breaking	1.73	1.73	2.10	1.93
Timbering (extraction drifts only)	3.20	3.20	-	3.20
Shoveling	3.27	3.27	-	3.27
Total labor	8.20	8.20	2.10	8.40
Feet per 8-hour shift	4.61	4.61	3.81	4.15
Power and supplies (per foot):				
Explosives (lbs. per foot)	5.50	5.50	4.32	5.18
Timber (board feet, extraction drifts only)	137.8	137.8	-	137.8
Power and supplies, percentage of total cost	-	-	-	25.5

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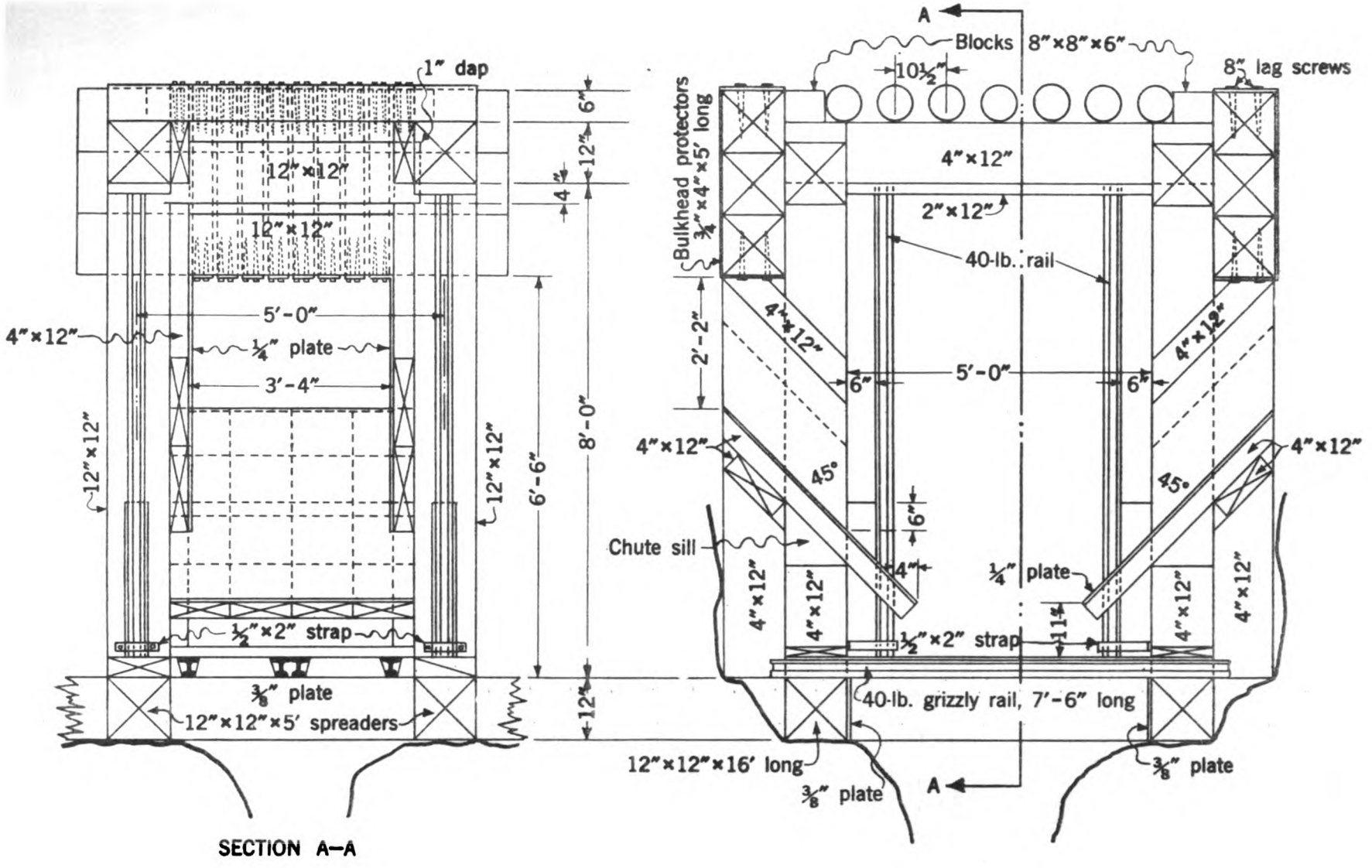

Figure 14. — Standard 2-way ore pass chute control

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